# STUDY OF SUPPORT SYSTEM AND STRATA BEHAVIOUR ANALYSIS AROUND UNDERGROUND WORKINGS IN COAL DEPOSITS

A THESIS SUBMITTED IN PARTIAL FULFILLMENT
OF THE REQUIREMENTS FOR THE DEGREE OF

**BACHELOR OF TECHNOLOGY**
**IN**
**MINING ENGINEERING**

BY

**ABHINAV SINGH BAGHEL**

NATIONAL INSTITUTE OF TECHNOLOGY
ROURKELA

**Department of Mining Engineering**
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Under the Guidance of
**Prof. S. Jayanthu**

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## CERTIFICATE

This is to certify that the thesis entitled, "***Study of Support System and Strata Behaviour Around Underground Workings in Coal Deposits***" submitted by Mr. Abhinav Singh Baghel in partial fulfillment of the requirement for the award of Bachelor of Technology Degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any University/Institute for the award of any Degree or Diploma.

Date – 13.05.2009

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An assemblage of this nature could never have been attempted without reference to and inspiration from the works of others whose details are mentioned in reference section. I acknowledge my indebtedness to all of them.

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# CONTENTS

## ABSTRACT

The estimation of vertical stress and its distribution over the underground mine workings is of prime importance. In this project work their estimation is been done with the help of numerical modeling by simulating the workings. Study and analysis of the stress distribution around development and depillaring workings in coal mines and vertical stress estimation is done.

The working has been modelled by writing a program code in FLAC5.0. The modelling is done in stages involving driving of galleries (development) to form three pillars and then the extraction of these pillars (depillaring) by slicing, then complete extraction to form ribs further followed by the judicious rob and burst of rib. The model is run in each of these stages to get the vertical stress distribution.

From the analysis of stress distribution through numerical model (FLAC5.0) for a depth cover of 95.5m following maximum vertical stress was observed as Maximum vertical stress over the pillar during development is about 3 MPa, Maximum vertical stress over the stook during development is about 4 Mpa and Maximum vertical stress over the rib during development is about 7 Mpa.

The outcome of the results show that the ultimate vertical stress increases considerably with increase in the depth cover and get concentrated over the area of excavation with high stress concentration over the pillars, stooks and ribs above the normal stress under the given depth cover.

## LIST OF FIGURES

# LIST OF TABLES

# 1. INTRODUCTION

Strata control deals with the adaptation of a system by which we could have a control on the strata movement to a desired level to make our workings safe and extraction of mineral possible. As the future of Indian mining lies in underground workings strata control is of prime importance.

## 1.1. STATUS OF COAL SEAMS IN INDIA

Nearly 61% of the total reserve of coal is estimated within 300m depth cover, distributed in all coalfields from Godavari Valley to Upper Assam. The prime quality coking coal of Jharia is available mainly in upper coal horizons while the superior quality non-coking coal of Raniganj is available in lower coal horizons. The quality coal of central India to Maharashtra is also available mainly in seams within this depth range. As a result all the mines worked such seams extensively, primarily developing on pillars and depillaring with sand stowing. With the unfavorable economics of sand stowing and non availability of virgin patches for further development, most of the mines have been working- splitting or slicing the pillars, winning roof or floor coals manually or with SDL, conveyor combination.

| State | Resource estimate as on 1.1.07 under depth | | | Total Reserve (Mt) |
|---|---|---|---|---|
| | 0-300m | 300-600m | 600-1200m | |
| A P | 7922 | 6514 | 3024 | 17461 |
| Chhattisgarh | 32167 | 8614 | 669 | 41450 |
| Jharkhand | 36998 | 14601 | 3285 | 54884** |
| **Jharia | -------14213--------------- | | 5217 | 19430 |
| Maharashtra | 6789 | 2698 | 183 | 9670 |
| M.P | 12902 | 6727 | 148 | 19777 |
| Orissa | 44636 | 16139 | 1224 | 61999 |
| W Bengal | 12361 | 10975 | 4999 | 28335 |
| Grand Total | 155785 | 80636 | 18749 | 255170 |
| % share | 61.24 | 31.66 | 7.35 | 100 |

**Table: 1: Depth wise coal resource estimate in various states of India as on 1st January, 2007**

The resource position of coal shows nearly 37% within 300-700m depth cover and a small portion (7 %) below 600m depth cover. Quality coal below 300m depth cover in Raniganj, Jharia, East and West Bokaro, North and South Karanpura, Sohagpur, etc should be the main targets for underground mining.

The country's coal production programme as envisaged in the 'Vision Coal – 2025' document is a quantum leap from the existing level of around 430.85 MT in 2007-08, being the terminal year of Xth 5 year plan, as can be seen from the table 2

Perpetually changing scenario due to unpredictable nature of geo-technical environment while mining minerals/coal makes mining one of the most hazardous peacetime occupations. This highly unpredictable and varying nature of working conditions in the mines exposes workpersons to dangerous conditions. Such conditions enjoin upon Indian state, mine operators, scientific mining institutions to take appropriate measures to reduce density of workpersons at potentially high risk zones i.e., moving front of drivages and depillaring workings, to reduce accidents. Limited potential of opencast reserves coupled with environmental considerations, land acquisition issues and availability of better grade coal at depth will renew the focus of coal industry to extract coal from deeper horizons by underground methods.

| **Producing Company** | **XI Plan (2011-12)** | **XII Plan (2016-17)** | **XIII Plan (2021-22)** | **XIV Plan (2025)** |
|---|---|---|---|---|
| | **(Production envisaged in M.T)** | | | |
| M/s CIL | 536 | 653 | 755 | 839 |
| Coal Equivalent CBM/UCG | - | 5 | 15 | 25 |
| M/s SCCL | 41 | 45 | 47 | 47 |
| Others | 44 | 75 | 125 | 175 |
| Grand Total | 621 | 778 | 942 | 1086 |

**Table 2. Production of coal envisaged in India**

## 1.2. STRATA CONTROL PROBLEMS

The strata control are to deal with proper management and methodology. In India a good system would result more safer mining atmosphere and high productivity. Geological discontinuities are a prime causative factor in strata-movement problems in underground collieries. Faults (normal, bedding, slips and slickensides) are the most important causative factors in the roof-fall index followed by bedding planes, joints and cleats. The presence of such discontinuities leads to roof instability, mainly because of poor cohesion/adhesion between them. These strata-movement problems are largely due to shear failure along normal faults, slips, slickensides, joints and cleats, whereas thinly bedded strata and bedding faults cave in because of tension failure. Strata-movement problems can be reduced by orienting roadways at 20-90 ~ to the direction of discontinuities, by planning narrower roadways and by augmenting the support density. Discontinuities, or defects in the roof, are the most dangerous geologic structures found in coal mine roof. Unseen breaks in otherwise solid roof may provide little warning of impending failure. Clay veins, slickensides, sandstone channels, and joints are the most common of such discontinuities. It is a rare coal mine that has never experienced some type of geologic roof disturbances. An understanding of the origin and occurrence of these features will greatly aid in their tracking and prediction. Support measures can be applied more appropriately if the roof damage and resulting loads are better understood.

Much benefit can be realized from careful and systematic roof fall analysis. Roof falls are the best exposures of roof, especially weak and defective roof. Uncovering the cause of roof falls can reveal trends, including regular shear patterns, weak bolt anchorages, damage from horizontal stress, bolt failures or poor bolt installations, overspanned intersections, and water swelling in clay fault gouge.

Features that lead to typical problems in underground coal mining include;

- Steeply dipping, faulted, folded, highly gassy beds under aquifers and protected land have remained virgin.
- Developed pillars under fires, surface features sterilized because of acute shortage of sand.
- Development has been in multi sections.

Highly stressed zones have been created due to barriers/stooks causing difficulty of undermining of the seams.

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Roof deterioration can range from a few inches of scale between bolts to complete failure in the form of a roof fall that could run for hundreds or thousands of feet. MSHA defines a reportable roof fall as any roof failure that (1) causes injury that has reasonable potential to cause death, (2) disrupts regular mining activity for more than an hour, (3) occurs at or above bolt anchorage, (4) impairs ventilation, or (5) impedes passage

Geological exploration to locate suitable panels for each set of equipment with seam thickness variation within the permissible limit, coal of quality and roof rock formation should be done in depth before introducing any such cost intensive technology with continuous miners in 300-400m depth cover and longwall technology below 400m depth cover. Necessary steps to ensure their success is summarized as follows,

- Shaft sinking technology should be perfected to develop access to deeper seams
- Back up facility – vertical and horizontal transport, processing and dispatch system should be compatible to the mass production technology.
- Equipment supply and spare availability should be ensured for efficient full life performance
- Man power preparation including training and on face operational skill should be developed on priority
- Work culture should be improved in respect of devotion, commitment and adaptation of modern technology with efficiency

# 2. LITRATURE REVIEW

*STRATA CONTROL*

At present, the most commonly practised pillar extraction method is the rib-and-slice depillaring. It consists of dividing a pillar in two or more stooks by driving a or two split roadway generally along the level and subsequently following the diagonal line of extraction, taking slices 3-5m wide while leaving an L-shaped rib (to be judiciously reduced) against the goaf and maintains the 'safe area of exposure' as per DGMS norms. Depillaring is thus carried out from dip to rise and from the panel (mine) boundary to its access. Timber supports like cogs and props are still an integral part of pillar extractions, though roof bolting is increasingly receiving attention not only because of paucity of timbers but also because of better reliability and efficiency.

## 2.1 FACTORS INFLUENCING PILLAR EXTRACTION

The extraction methods are employed with high mechanisation (i.e. the use of continuous miner) or with the intermediate mechanisation (with SDL or LHD) in order to achieve objectives of safety, productivity and competitiveness. The choice of an approach depends, by and large, the extent and behaviour of caving, which primarily is governed by the following factors, detailed elsewhere[1-4]:

- panel geometry and depth of cover
- geotechnical issues and geology
- coal extraction processes and sequence of mining
- management control issues.

It may be noted that pillar extraction environments has inherent variations, complex caving behaviours associated, compared to the environments of longwalling, because of [5-8]:

- transient and rapid changes occurring due to planned and unplanned stooks (fenders) and remnants,
- irregular goaf geometry,
- coal extraction processes and

- sequence of mining

It is not possible to incorporate above factors partially or in totality in any formulation or thumb rule developed so as to estimate support requirements in depillaring areas. For this, we need to resort to three-dimensional numerical modelling taking in-situ pre-excavation stresses and other physico-mechanical properties as inputs to the simulated models like the case-studies discussed latter in this paper as separate sub-titles. Based on the research studies conducted under a Mininstry of Coal S&T grant-in-aid project and the extensive numerical modelling excersies by the second author, the following formulation are recommended, as detailed elsewhere [CIMFR Report , 2007]:

**For slice junction,**
$$SLD_{jn} = \frac{\gamma . H^{0.50} . K^{0.64} . W^{1.17}}{R^{0.90}}$$

**Within slice,**
$$SLD_{sl} = \frac{\gamma . H^{0.67} . K^{0.84} . W^{1.74}}{R^{1.42}}$$

**In the split gallery,**
$$SLD_{sp} = \frac{\gamma . H^{0.52} . K^{0.59} . W^{1.12}}{R^{1.02}}$$

**For goaf edge,**
$$SLD_{ge} = \frac{\gamma . H^{0.54} K^{0.49} . W^{0.89}}{R^{0.79}}$$

where, $\gamma$ is the weighted average rock density of the immediate roof strata, t/m$^3$, H is depth of cover, m, K is the ratio of horizontal to vertical in situ stress, W is the width of split or slice, m and R is the weighted average RMR of the immediate roof rock. $SLD_{jn}$, $SLD_{sl}$, $SLD_{sp}$ and $SLD_{ge}$ are the required support density in t/m$^2$ at the slice junction, within slice, in the split gallery and at the goaf edge respectively.

It is estimated by mining-experts that large reserves, more than 2500MT of mineable coal, equivalent to 7-8 years production in India are locked-up in developed bord and pillar workings, including multiple and thick seams. Extraction of these developed pillars is a challenge to the mining community. Extraction with the use of timber-supports will not be technically and financially viable in this case also.

## 2.2 ROCK REINFORCEMENT

Rock mass contains geological discontinuities/weakness and hence has the strength-parameters proportionately less than ‘intact rock’, the latter is defined as the rock-portion between the any two adjacent geological discontinuities and thus generally devoid of any such weakness. Rock reinforcement is a specific technique of rock improvement, which includes all techniques, which seek to increase the strength and decrease the deformability characteristics of a rock mass. The prime objective of rock reinforcement is to improve the shear and tensile strength of the rock mass adjacent to underground excavations.

Reinforcement terminology includes description of the reinforcing elements, installation procedures and the philosophy behind the reinforcement scheme design. For example, some of this terminology includes reinforcing elements (anchors, dowles, bolts, pins, nails, cables, tendons), installation procedures (pre and post-inforcement, pre and post tensioning, grouted and ungrouted, bonded and debonded, coupled and uncoupled, permanent and temporary reinforcement) and reinforcement scheme philosophy (strata reinforcement, rock support, cable doweling, rock anchoring, pattern of reinforcement and spot bolting).

There are factors related to installation, which can optimise the load transfer and the performance of the reinforcing element in response to rock mass behaviour;. These include, among others, the life of installation, the timing of installation and the provision of initial tension in the reinforcement and procedures for semi-permanent or permanent excavations. In many applications it has been found that there are substantial benefits in safety and productivity associated with pre-reinforcement of excavations. Pre-reinforcement can prevent premature failure of the rock and provides a safer working condition for the installation of further reinforcement or support. Some reduction in overall reinforcement requirements are sometimes possible through post-reinforcement or reinforcement  at an appropriate time after the creation of the excavation.

Similarly, sometimes it is desirable to provide the reinforcing element with an initial pre-tension. Post tensioning is the tensioning or re-tensioning of devices subsequent to installation. Further tension may develop with time as the rock mass moves due to subsequent excavation activity, stress changes or creep. This possibility must be explored and allowed for to avoid subsequent overstressing and rupture.

**Rock load**

Maximum load (P) that is required to be supported in the split and slice can be estimated using the following formula and as detailed elsewhere [Kushwaha, 2005]:

$$P = \gamma .\ SF_{1.5h} \qquad (8)$$

where, $\gamma$ = weighted average rock density, 2.5 t/m$^3$ (carbonaceous shale)

$SF_{1.5h}$ = height of safety factor contour up to 1.5 in the roof strata in the simulated model.

## 2.3 SUPPORT ESTIMATION

A pattern of support may be proposed using the following formula such that an adequate support safety factor (about 1.1-1.25 in depillaring areas, about 1.5-2.0 for permanent roadways) is achieved:

$$S = \frac{n\ x\ b_c}{w\ x\ sp}$$

where, n =the number of bolts/props in a row

$b_c$= fully column grouted roof bolt capacity, 8 tonne

fully column resin roof bolt capacity, 16 tonne

capacity of timber props, 10 tonne

capacity of timber cogs, 20 tonne

w =width of the slice, here 4.2m

sp= spacing between two rows

Support safety factor = **S / P**

## 2.4 GUIDELINES FOR DRAWING OF SUPPORT PLANS IN BORD & PILLAR WORKINGS IN COAL MINES

**General:**

The various stages of designing a suitable support system and ensuring successful installation are basically as follows:

(a) A geotechnical survey and interpretation of survey findings

(b) Selection/designing of support system based on above interpretation

(c) Selection of equipment

(d) Actual installation process and

(e) Monitoring of the system.

**Two systems are particularly used to characterize mining ground conditions.**

1. **Barton's Q-system ( Rock quality index, Norwegian Geotechnical Institute)**

**It is evaluated as**

$$Q = \frac{RQD \times j_r \times j_w}{j_n \times j_a \times SRF}$$

Where *RQD* = rock quality designation

$J_n$ = joint set number

$j_r$ = joint roughness number

$j_a$ = joint alteration number

$j_w$ = joint water reduction number

and *SRF* = stress reduction factor.

Based on the value of Q the rock mass can be described as "exceptionally good" (Q=400 to 1000) to "exceptionally poor" (Q=0.001 to 0.01 ). Using the Q value, the maximum unsupported span of roof can be estimated by the formula:

Span (m) = 2 x ESR x Q $^{0.4}$

Where ESR is excavation support ratio (which is 3 to 5 for temporary mine workings and 1.6 for permanent workings). The rock load ($P_{roof}$) can be estimated from the empirical formula:

$$P_{roof}\ (t/m^2) = \frac{2.0 \times F}{Jr \times Q^{0.33}}$$

Where F = 1 if Jr is 9 or more

Or F= ((Jr $^{0.5}$)/3 if Jr is less than 9.

Depending on the different values of the parameters and Q, 38 support categories have been identified.

2. **Bieniawski's RMR system**

There are five parameters in this classification:

(i) Intact rock strength

(ii) RQD

(iii) Joint spacing

(iv) Condition of joints

(v) Ground water seepage

Rating division for each of the parameters is given and RMR is sum of five ratings. Based on RMR, the roof is classified as very good (RMR:80-100) to very poor (RMR:0-20). From this estimation of rock load is derived using theoretical relation and support guide is provided.

**Suitability of Q-system/ RMR system**

These two classifications have been applied to about 30 Indian coal mines. The Q classification is suitable for highly jointed rocks for hard rock conditions . Most of the parameters in this system are based on joint attributes whereas stability in coal mines is not merely joint controlled. The SRF has no relation with the stress field occurring around multiple openings like coal mine roadways. The parameter description in Q system leave much to subjective judgement.

The RMR system gives results nearer to actual roof conditions . t was recognized that in the most of the Indian coal mines, bedding planes, structural features and weathering of roof rocks are then major causes of roof failure. In Bieniawski's approach, consideration is not given to sedimentary features, structural features other than joints and weatherability of rocks. Deviation in the results also arise from the weightages for the parameters which need to be adjusted to Indian rock conditions.

## 2.5 CMRI-ISM ROCK MASS CLASSIFICATION

This rock mass classification system is being used regularly by academic and research institutes. The five parameters used in the classification system and their relative ratings are summarized below:

| S. No. | Parameter | Max. rating |
|---|---|---|
| 1 | Layer thickness | 30 |
| 2 | Structural features | 25 |
| 3 | Rock weatherability | 20 |
| 4 | Strength of roof rock | 15 |
| 5 | Ground water seepage | 10 |

**Table 3. CMRI-ISM prescribed parameters for RMR determination**

The five parameters should be determined individually for all the rock types in the roof upto a height of at least 2 m.

1. **Laying thickness:** Spacing between the bedding planes or planes of discontinuities should be measured using borehole stratascope in a e m long drill made in the roof. Alternately, all bedding planes or weak planes within the roof strata can be measured in any roof exposure like a roof fault area, shaft section or cross measure drift. Core drilling shall be attempted wherever feasible and the core log can be used to evaluate RQD and layer thickness. Average of five values should be taken and layer thickness should be expressed in cm.

2. **Structural Features:** Random geological mapping should be carried out and all the geological features (discontinuities like joints, faults and slips, and sedimentary features like

cross bedding, sandstone channels) should be carefully recorded. The relative orientation, spacing and degree of abundance for all these features shall be noted. Their influence on gallery stability should be assessed and the structural index for each feature should be determined from the Table 1 as given below.

3. **Weatherability** : ISRM standard slake durability test should be conducted on fresh samples from the mine to determine the susceptibility of rocks to weathering failure on contact with water or the atmospheric moisture. For this test, weigh exactly any ten irregular pieces of the sample ( the total weight should be between 450- 500 g); place them in the test drum immersed in water and rotate it for 10 min at 20 rpm; dry the material retained in the drum

after the test and weigh it again. Weight percentage of material remaining after test is the final slake durability index, expressed in percentage. Mean of three such first cycle values should be taken. Core may be broken to obtain the samples.

4. **Rock Strength:** Point load test is the standard index text for measuring the strength of rocks in the field. Irregular samples having ratio of 2:1 for longer axis to shorter axis can be sued for the test. The sample is kept between the pointed platens and the load is applied gently but steadily. The load at failure in kg divided by the square of the distance between the platens in cm gives the point load index ($I_s$). The mean of the highest five values out of at least 10 sample tests should be taken. The compressive strength of the rocks can be obtained from the irregular lump point load index for Indian coal measure rocks by the relation:

**$Co = 14\ I_s$ ( in $kg/cm^2$)**

5. **Ground water:** A 2m long vertical borehole should be drilled in the immediate roof and the water seeping through the hole after half an hour should be collected in a measuring cylinder. The average of three values from three different holes should be taken and expressed in ml/minute.

***Rock Mass Rating(RMR) is the sum of five parameter ratings. If there are more than one rock type in the roof, RMR is evaluated separately for each rock type and the combined RMR is obtained as:***

$$\textbf{Combined RMR} = \frac{\sum (\text{RMR of each bed} \times \text{bed thickness})}{}$$

$\sum$ **(Thickness of each bed)**

The RMR so obtained may be adjusted if necessary to take account for some special situations in the mine like depth, stress, method of work

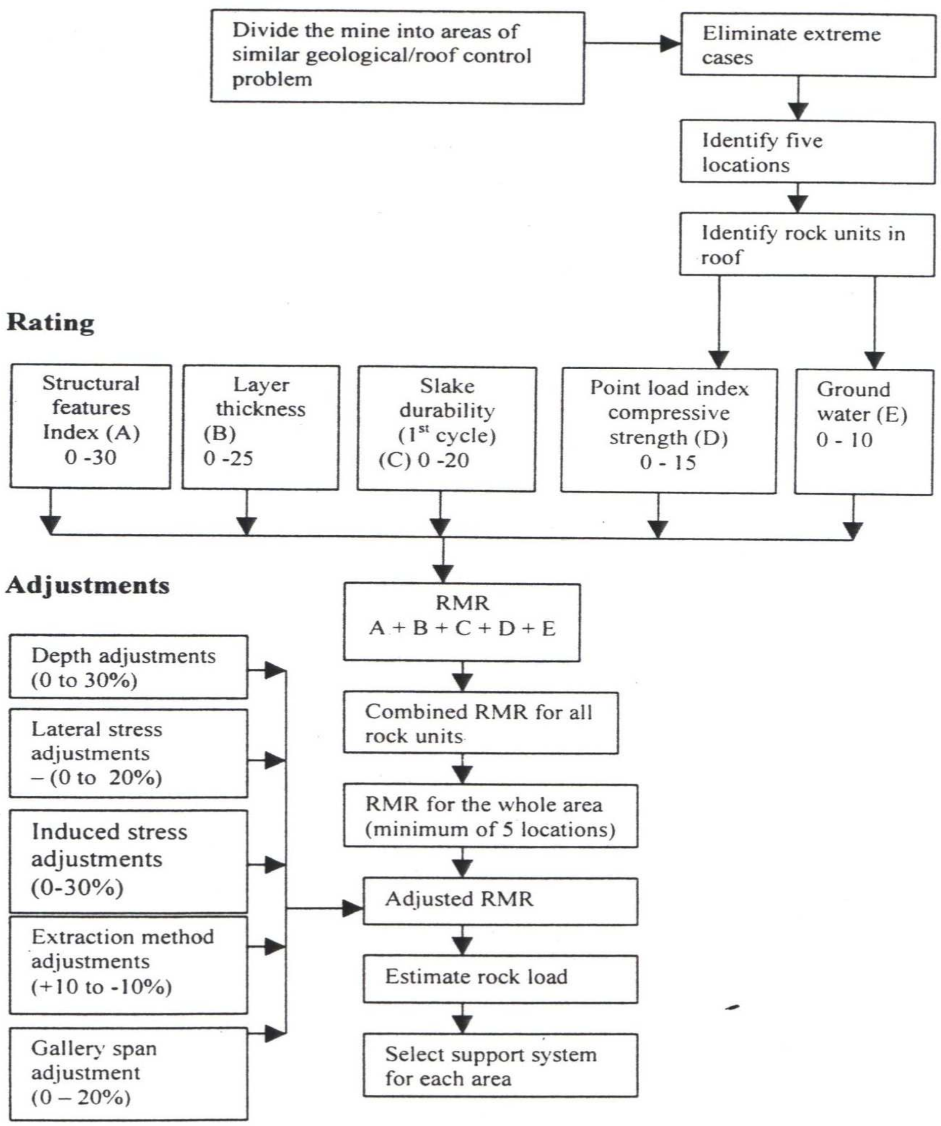



**FIG.1 Flow-sheet for deriving RMR**

## 2.6 DESIGN OF SUPPORT FOR DEPILLARING WORKINGs

In general, Rock Mass Rating (RMR) is used for design of supports in development galleries. However, due to limitations of its application to depillaring workings, many investigators adopted various approaches such as Q-classification of rock mass, numerical modeling etc for design of support system in depillaring workings, Some times, it is also required to design support in a depillaring panel having widely varying geomining conditions with different support density.

For the purpose of support design in a typical depillaring area, Barton's Rock mass classification index- Q was also determined as follows:

Q = {RQD/Jn} {Jr/Ja} {Jw/SRF} ----(1)

Rock Quality Designation = f (layer thickness) = 97

Jn = no joints were observed in the roof = 4 for galleries

= 12 for junctions

= 20 for goaf edges

Ja = Plant impressions are frequent in the roof; however kettle bottoms/sandstone channels/slickensides are not perceptible = 1

Jw = generally dry up to 8 ml of water per minute seepage.= 1

Jr = Smooth planar joints = 1

SRF values for various geometries during depillaring are as follows:

| | c/Ms | SRF |
|---|---|---|
| *For galleries and junctions*: | >10 | 1 |
| | 1 - 10 | 1-2 |
| *For slices:* | >5 | 2 |
| | 2.5 - 5 | 3 - 5 |
| | <2.5 | 5 |
| *For goaf edges :* | any value | 10 |

Roof pressure could be estimated by the relations based on the Q value adjusted to the geometrical conditions:

For joint set number (Jn)> 9, the roof pressure(Proof) = 2/Jr x (5Q)-1/3 --(2)

For Jn < 9, Proof = 2/3 Jn1/2 /Jr x (5Q)-1/3 --(3)

## 2.7 DESIGN OF SUPPORT FOR LONGWALL WORKINGS

Following approaches are generally used for estimation of the support requirement for longwall panels. The support requirement based on these approaches along with the adequacy of the supports in a typical panel is as follows:

**1. Bary's Approach :**

According to Bary's method {1969}, the maximum support capacity (in tonnes) is calculated by the following equation :

$$\mathbf{P = s\ L\ r\ h} \qquad \text{---} \qquad \mathbf{(1)}$$

$$\mathbf{L = e + b + m}$$

$$\mathbf{h = M/(K\text{-}1)}$$

where,

| | |
|---|---|
| s = support spacing, m | = 1.5 m |
| L = total length of overhang, m | = 14.605 & 19.605 m |
| e = length of overhang behind canopy ,m | = 10 & 15 m |
| b = distance between goaf edge of canopy and face after cut, m | = 4.605 m |
| m = extraction thickness, m | = 0.8 m |
| r = density of roof rock | = 2.5 g/cc, |
| h = caving height, m | |
| M = height of extraction, m | = 4.5 m |
| K = bulking factor | = 1.06 and 1.1 |

Details of estimation of support resistance on the basis of the above approach, using bulking factor of 1.06 and 1.1, are as follows :

| Caving Height, m | Overhang (e), m | Load, t |
|---|---|---|
| 45 | 10 | 2465 |
| 75 | 10 | 4108 |
| 45 | 15 | 3308 |
| 75 | 15 | 5514 |

During heavy loading in a typical situation, the leg pressure crossed yield pressure with about 1.8-2.2 m leg closure. The load on support were at least four times the support capacity, i.e., 2465 ton with about 45 m caving height and 10 m overhang.

**2 Sigott's Approach :**

According to Sigott's approach {Habenicht, 1972}, minimum required support resistance can be calculated as follows :

**P = (3/4)a ($g^2$/n) b r M/(K-1), t/linear meter --- (7)**

where,

a = reduction factor = 0.9,

g = overhang factor = (b+e)/b,

b = distance from goaf-edge to face after cut, m = 4.605 m

e = overhang, m = 10 & 15 m

n = number of chocks per linear meter = 1/1.5

M = extraction height, m = 4.5 m

r = roof rock density, gm/cc = 2.5

K = bulking factor = 1.06 and 1.1

| **Overhang (m)** | **Load / support * (t)** | |
|---|---|---|
| | For K = 1.06 | For K = 1.1 |
| 10 | 8794 | 5276 |
| 15 | 15845 | 9507 |

**As per the above approach, assuming a bulking factor of 1.1, it is concluded that the minimum support resistance should be about 4285 t.**

**3 Siska's Approach :**

The method of estimation of support density with strong sandstones is suggested by Siska {1972}

$$\mathbf{R = r*M*K_{OZ}*K_s*\{1/(K-1)\}*K_z}$$

where,

R = support density in $t/m^2$ after cut

M = height of extraction, m = 4.5 m

$\gamma$ = roof rock density, g/cc = 2.5 g/cc

K = bulking factor = 1.06 and 1.1

$K_{OZ}$ = coefficient of delayed caving = $(V_1+V_2)/V_1$

$V_1$ = volume of the rock immediately over the support (taken for 45 and 75 m caving heights)

$V_2$ = increase in volume due to delayed caving (taken for 10 and 15 m overhang)

$K_s$ = coefficient of self-supporting ability of the overlying strata = $(V_3+V_4)/V_3$

$V_3$ = volume of the rock over the support (4.605 x 1.5 x 1.2 $m^3$)

$V_4$ = volume of rock self supporting (considered as '0' for the 0.3 m thick shale and 0.9 m thick coal beds in the immediate roof)

$K_z$ = coefficient of influence of support in the waste; for caving the value is 1

| **Overhang (m)** | **Support density ($t/m^2$)** | |
|---|---|---|
| | For K = 1.06 | For K = 1.1 |
| 10 | 594 | 357 |
| 15 | 797 | 478 |

**From the above, the required support capacity of chocks would be in the range of 534 to 1194 $t/m^2$, with a safety factor of 1.5.**

## 4 Das' Approach

This approach is based on Indian longwall mining experiences, where caving is predominantly parting-plane controlled. The capacity of the powered support in tonne ($T_y$) is estimated as (Das and Ghose, 1996) :

$$\mathbf{T_y = (R \times Sf)/(\eta i \times \eta 1)}$$

where,

R = Resistance to be offered by the powered supports in tonnes

Sf = Factor of safety,

=1.5 in case of medium strength to stronger massive sandstone

=1.2 in other cases

$\eta_i$ = efficiency factor of the powered support due to inclination of legs

= 0.85 for chock shield support

= 0.8 for shield support

$\eta_1$= efficiency factor of the powered support due to leakage in the pipelines/valve system and mechanical defects

= 0.9

$$\mathbf{R = [\{W1\,(L_1/2) + (H_1 \,Tan\, \alpha_1/2) + W_2\ \{(L_1 + H_1\, Tan\, \alpha_1/2 + (H_2\, Tan\, \alpha_2)/2)\} + 3/8\ W_3\ \{1/3(2L_1 + H_3\, Tan\, \alpha_3) + H_1\, Tan\, \alpha_1\, H_2\, Tan\, \alpha_2\}]/P}$$

where,

$W_1$, $W_2$, $w_3$ are the weight of 1st, 2nd and 3rd roof layers

$H_1$, $H_2$, $H_3$ are the thickness of 1st, 2nd and 3rd roof layer

$L_1$ = distance between longwall face and caving edge

$90^o - \alpha_1$, $90^o - \alpha_2$, $90^o - \alpha_3$ = caving angle of the 1st, 2nd and 3rd roof layers

P = distance from face to the centroid of resistance offered by the powered support.= 3.78 m

## 5 Josien's Approach

Josien and Gouilloux (1978) found the relationship correlating convergence at a face with load bearing capacity of the powered supports;

$$\mathbf{Cv = (ah)^{0.75}\, D^{-1/4}\, ((6800/R)+ 66)}$$

where,
Cv = Average convergence, mm/m of face advance
A = Subsidence factor, for caving=1
D = Depth of the mine , m= 180 m
h = Extraction height, m= 4.5 m
R = Load bearing capacity of the support, t per meter of the face

It seems that its application is again limited to convergence values exceeding 55 mm/m, as undesirable negative values would be associated with R if Cv < 55 mm/m. As per the above equation, with Cv =60 mm/m, the support capacity required would be 1142 t/m of face. This is equivalent to the required load bearing capacity of about 1713 tons per shield.

**As per this approach also, even for a height of extraction of 3 m, the support capacity should be more than 900 ton per shield with a safety factor of 1.5 for better strata control. Obviously, the capacities required would be still higher in case of greater extraction thickness.**

## 6 Gupta's Approach :

For better face conditions, the leg closure should be within 3 mm per m of face advance (Gupta, 1982; Gupta and Ghose, 1992; Gupta and Farmer, 1985). And the mean load density (MLD) should have been as follows for Kottadih :

$$\mathbf{C = K\, (M/1.8)^{0.75}\, Exp^{-4.59MSLD}}$$

where,
C = face convergence, which should be < 3 mm/m of face advance to achieve good roof and face conditions
K is a constant= 40 for competent roof

= 150 for easily caving roof

M = extraction height, m

MSLD = mean setting load density ($t/m^2$)

(MLD-MSLD) = 0.224 + 0.438 MSLD

For K = 100, MSLD = 91.36 $t/m^2$,

and mean support density (MLD) = 131 $t/m^2$

As per this approach, the mean load support density should be more than 130 $t/m^2$ for better strata control.

**2.8 MONITORING AND CONTROL OF STRATA MOVEMENT:**

CONVENTIONAL BORD AND PILLAR EXTRACTION

- To minimize the dangers from weighting on the pillar due to overhanging of roof in the goaf and to ensure that as small an area of un-collapsed roof as possible is allowed in the goaf, a suitable code of practice for induced blasting shall be evolved in consultation with a scientific organization keeping in view the depth of induce shot holes being not less than 2.7 m, direction & spacing of shotholes, explosives used etc. so as to limit the rate of convergence [i.e., the ratio of C1/C2 is equal or less than 2, where C1 is daily convergence at a site in a day "n" and C2 is the average daily convergence at the site upto the previous day i.e. day (n-1)] and also to ensure complete filling of the goaf and release of any abutment pressures.

- Convergence recording stations shall be installed at all junctions situated within two pillar distance from pillar under extraction in the proposed panel. Monitoring of readings at convergence recording stations shall be done in every shift by a competent person duly authorized by the manager and the measurements shall be recorded in a bound paged book and the same shall be counter signed daily by the Under Manager of the shift and Asst. Manager in charge. All the work persons shall be withdrawn from the abutment zone if the ration of C1/C2 is equal to or more than 2 as given above and steps shall be taken to release the goaf abutment pressure by induced blasting. The Safety Officer shall co-ordinate recording, analysis and interpretation of the readings and advise the Officers/ Officials daily at the mine.

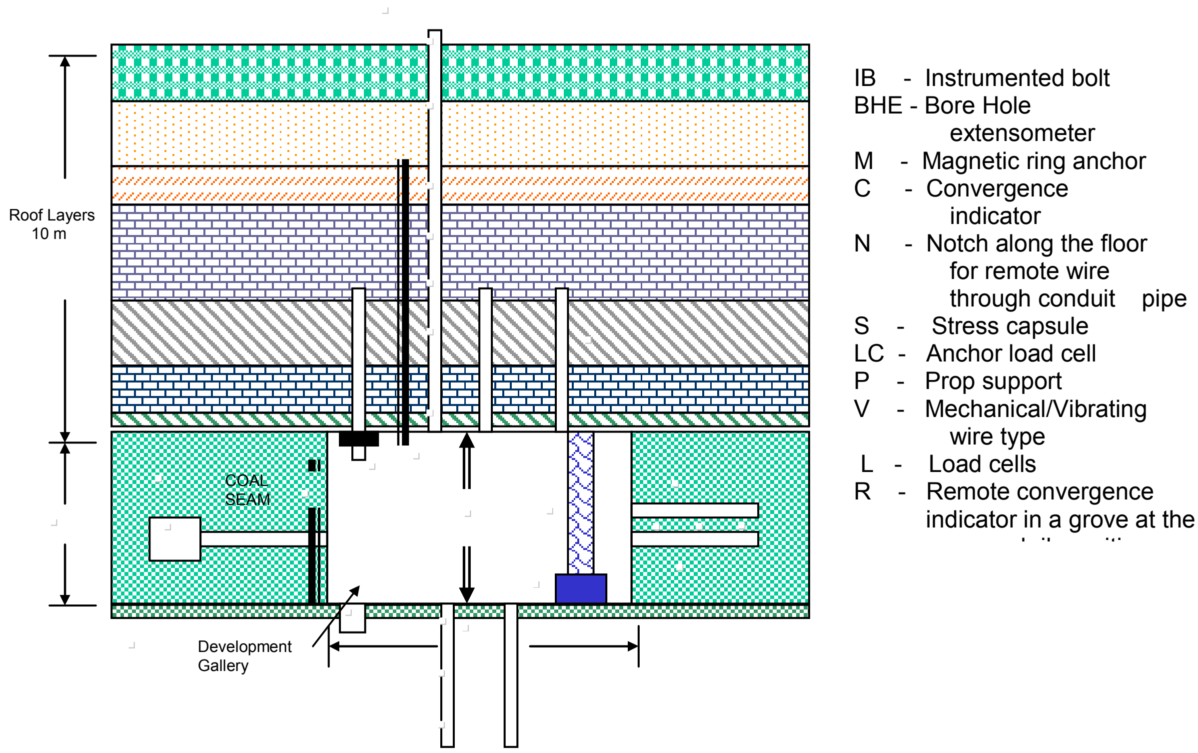



**FIG.2 TYPICAL INSTRUMENTS FOR STRATA MONITORING**

## 3. CASE STUDY OF R – 6 MINE NCPH COLLIERY, SECL

PARTICULARS ABOUT THE SEAM

1. Name of seam – No.3
2. thickness – 5.6 m
3. Thickness of seam section proposed to be depillared – full thickness
4. Dip – 1 in 43

PARTICULARS ABOUT WORKINGS

1. Max and Min height of working - panel no. 35 – 240m * 119m
2. Max and Min size of pillar - 39.5 m* 39.5 m
3. location of horizon - development is done along floor 2.6 m of coal had been left in the roof
4. age of working - 39 yrs
5. percentage of sand stone over the proposed panel – 58 % Dolorite sill 79.22 m thickness lies over the proposed panel just 0.6 m below the surface
6. gas emission - Degree – I

There is a prominent problem of the caving characteristic as the dolorite sill make it difficult to predict. The sandstone above the coal appears to be competent and not fractured. It is therefore proposed to support the coal and mid seam parting by anchoring long cables into the sandstone.

POINT ANCHORED CABLES

It is assumed that the sand stone is a good anchorage location and that the maximum thickness of failed coal and shale is 4 m.

The cables should have the strength of 50 tonnes, and length need to be 6 m. the anchorage on the sandstone must must be secure and the end fitting must be of the same strength as the cable.

Also the minimum length of resin encapsulation should be 2 m.in addition the cables must be connected via a heavy duty strap( 250 mm wide and 6mm thick steel).

It could be installed using rockbolt drilling methods and anchored using standard rockbolts resins. Nonetheless additional pairs of bolt , 2.4 m long and 22 mm dia full column resin bonded bolts may be installed along discontinuities.

# 4. NUMERICAL MODEL

A **computer simulation**, a **computer model** or a **computational model** is a computer program, or network of computers, that attempts to simulate an abstract model of a particular system. Models can take many forms, including but not limited to dynamical systems, statistical models, differential equations, or game theoretic models.

Often when engineers analyze a system to be controlled or optimized, they use a mathematical model. In analysis, engineers can build a descriptive model of the system as a hypothesis of how the system could work, or try to estimate how an unforeseeable event could affect the system. Similarly, in control of a system, engineers can try out different control approaches in simulations.

A mathematical model usually describes a system by a set of variables and a set of equations that establish relationships between the variables. The values of the variables can be practically anything; real or integer numbers, boolean values or strings, for example. The variables represent some properties of the system, for example, measured system outputs often in the form of signals, timing data, contours, and event occurrence (yes/no). The actual model is the set of functions that describe the relations between the different variables. Here FLAC ( Fast Lagrangian Analysis of Continua )5.0 has been used for simulation and analysis.

## 4.1 FLAC 5.0

*FLAC* is a two-dimensional explicit finite difference program for engineering mechanics computation. This program simulates the behavior of structures built of soil, rock or other materials that may undergo plastic flow when their yield limits are reached. Materials are represented by elements, or zones, which form a grid that is adjusted by the user to fit the shape of the object to be modeled. Each element behaves according to a prescribed linear or nonlinear stress/strain law in response to the applied forces or boundary restraints. The material can yield and flow and the grid can deform (in large-strain mode) and move with the material that is represented. The explicit, Lagrangian calculation scheme and the mixed-discretization zoning technique used in *FLAC* ensure that plastic collapse and flow are modeled very accurately. Because no matrices are formed, large two-dimensional calculations can be made without excessive memory requirements. The drawbacks of the explicit formulation (i.e., small timestep limitation and the question of required damping) are overcome to some extent by automatic inertia scaling and automatic damping that do

not influence the mode of failure.

Though *FLAC* was originally developed for geotechnical and mining engineers, the program offers a wide range of capabilities to solve complex problems in mechanics. Several built-in constitutive models that permit the simulation of highly nonlinear, irreversible response representative of geologic, or similar, materials are available.

However, it offers several advantages when applied to engineering problems.

1. The input language is based upon recognizable word commands that allow you to identify the application of each command easily and in a logical fashion (e.g., the **APPLY** command applies boundary conditions to the model).
2. Engineering simulations usually consist of a lengthy sequence of operations — e.g., establish in-situ stress, apply loads, excavate tunnel, install support, and so on. A series of input commands (from a file or from the keyboard) corresponds closely with the physical sequence that it represents.
3. A *FLAC* data file can easily be modified with a text editor. Several data files can be linked to run a number of *FLAC* analyses in sequence. This is ideal for performing parameter sensitivity studies.
4. Theword-oriented input files provide an excellent means to keep a documented record of the analyses performed for an engineering study. Often, it is convenient to include these files as an appendix to the engineering report for the purpose of quality assurance.
5. The command-driven structure allows you to develop pre- and post-processing programs to manipulate *FLAC* input/output as desired. For example, you may wish to write a mesh-generation function to create a special grid shape for a series of *FLAC* simulations. This can readily be accomplished with the *FISH* programming language, and incorporated directly in the input data file.

## 5. STRATA BEHAVIOUR STUDIES

It involves the analysis of the distribution of vertical stresses along with their numerical values, around the workings during stages of development and depillaring in the coal seam, by using the simulation technique of FLAC5.0.

## 5.1 PARTICULARS ABOUT SEAM

- Seam thickness – 7.5 m
- Pillar size – 25 m
- Gallery size – 4.8 m × 3 m
- Width of split – 5 m
- Rib thickness – 2.5 m
- Depth cover – 95.5 m

## 5.2 GEO-MECHANICAL PROPERTIES

The geo-mechanical properties taken for the different zones under analysis are listed as follows:

*COAL SEAM (7.5m, 95.5-103m depth)*

- Elastic shear modulus – $2.2 \times 10^9$ Pa
- Elastic bulk modulus – $3.67 \times 10^9$ Pa
- Tension limit – $1.86 \times 10^6$ Pa
- Density- 1427 $Kg/m^3$
- Cohesion – $1.85 \times 10^6$ Pa
- Angle of friction - 30°

*SANDSTONE ROOF & FLOOR OF SEAM (0-89.5m and 103-203 depth)*

- Elastic shear modulus – $4 \times 10^9$ Pa
- Elastic bulk modulus – $6.67 \times 10^9$ Pa
- Tension limit – $9 \times 10^6$ Pa
- Density - 2300 $Kg/m^3$

- Cohesion - $12\times10^6$ Pa
- Angle of friction - 45°

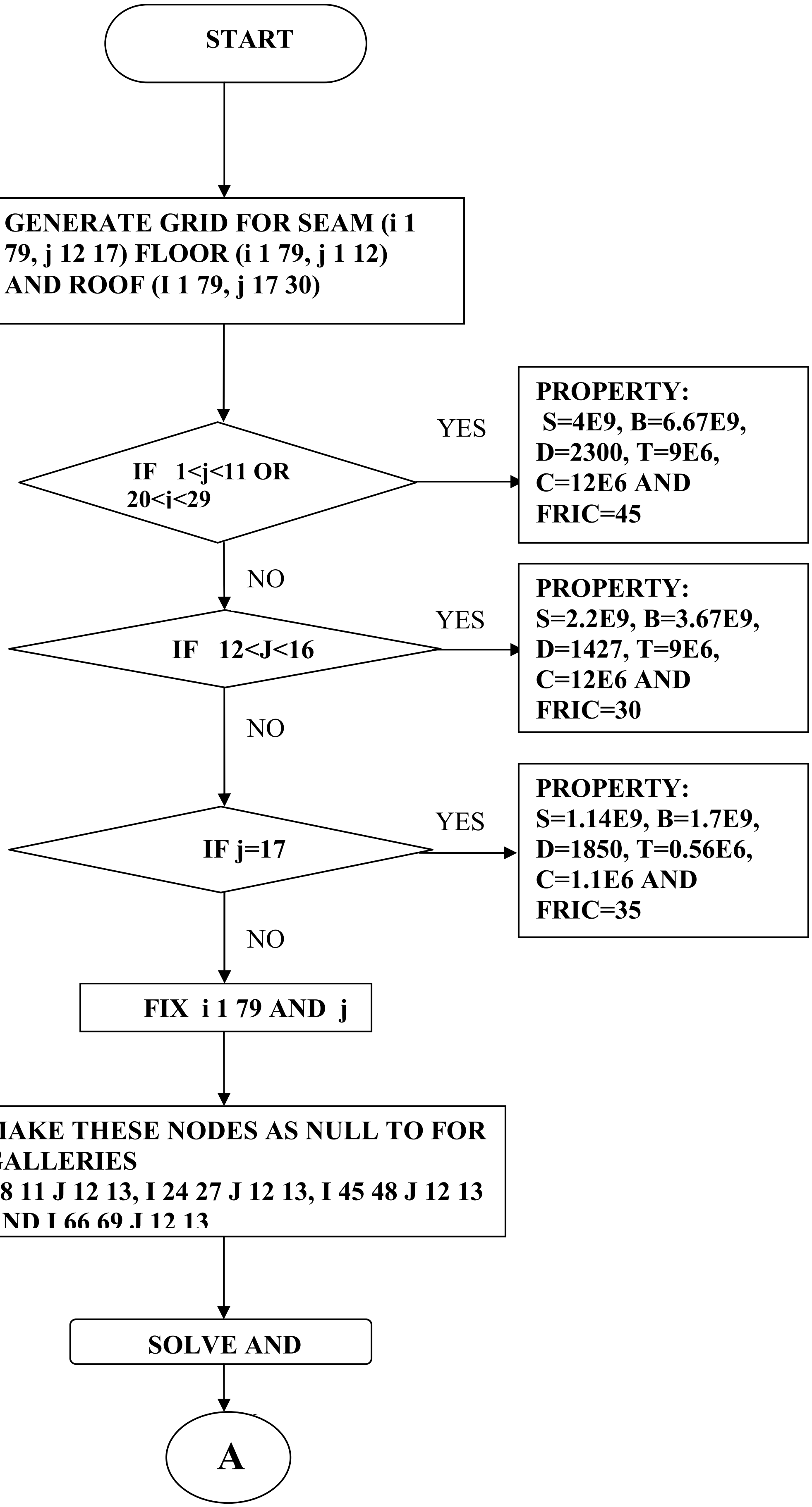
*IMMIDIATE ROOF (6m, 89.5-95.5m depth)*

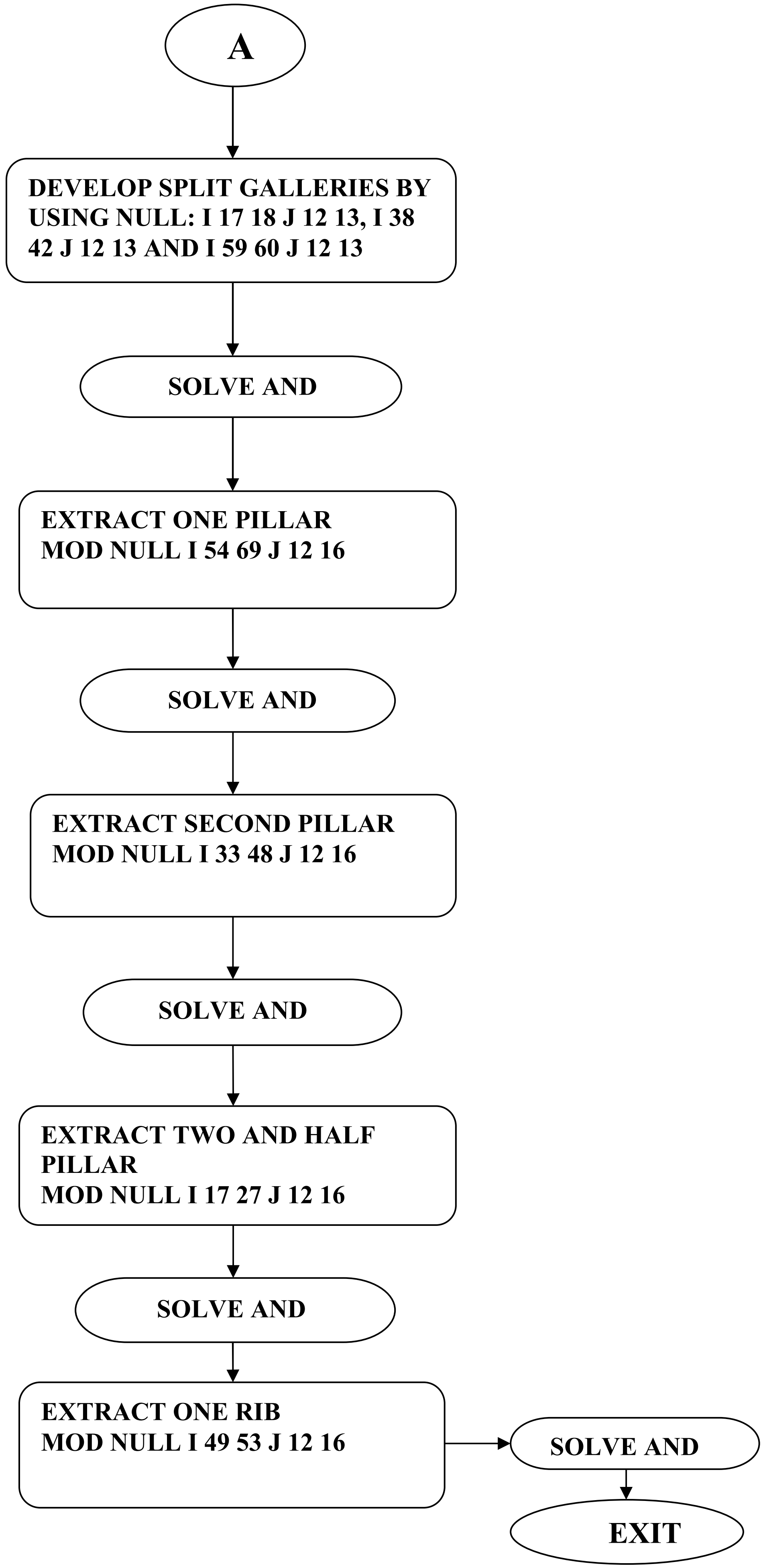
- Elastic shear modulus – $1.14\times10^9$ Pa
- Elastic bulk modulus – $1.7\times10^9$ Pa
- Tension limit – $0.56\times10^6$ Pa
- Density- 1850 $Kg/m^3$
- Cohesion – $1.1\times10^6$ Pa
- Angle of friction - 35°

## 5.3 MODELLING OF WORKINGS

The working has been modelled by writing a program code in FLAC5.0. The modelling is done in stages involving driving of galleries (development) to form three pillars and then the extraction of these pillars (depillaring) by slicing, then complete extraction to form ribs further followed by the judicious rob and burst of rib. The model is run in each of these stages to get the vertical stress distribution.

**FIG3.** **FLOW CHART OF THE ALGORITHM OF PROGRAM CODE**




A
DEVELOP SPLIT GALLERIES BY USING NULL: I 17 18 J 12 13, I 38 42 J 12 13 AND I 59 60 J 12 13
SOLVE AND
EXTRACT ONE PILLAR
MOD NULL I 54 69 J 12 16
SOLVE AND
EXTRACT SECOND PILLAR
MOD NULL I 33 48 J 12 16
SOLVE AND
EXTRACT TWO AND HALF PILLAR
MOD NULL I 17 27 J 12 16
SOLVE AND
EXTRACT ONE RIB
MOD NULL I 49 53 J 12 16
SOLVE AND
EXIT

# 6. ANALYSIS AND RESULTS

The 3 pillars have been modelled using FLAC5.0 with 4 galleries. The vertical stress distribution was observed during different stages as:

Development of pillars:-

The vertical stress distribution over the center of pillar, near corners of pillar and at the sides of pillar are found to be $5\times10^5$ Pa, $2\times10^6$ Pa and $3\times10^6$ Pa respectively.

The stress is distributed in high concentration at the sides of pillars.

Splitting of pillars:-

The vertical stress distribution over the center of stooks , near corners of stooks and at the sides of stooks are found to be $5\times10^5$ Pa, $3\times10^6$ Pa and $4\times10^6$ Pa respectively.

With the splitting the stress concentration increases.

After the extraction of 1 pillar:-

The vertical stress distribution over the goaf and at the goaf edges are found to be $1\times10^6$ Pa and $6\times10^6$ Pa respectively.

The high value of stress is generated at the goaf edge.

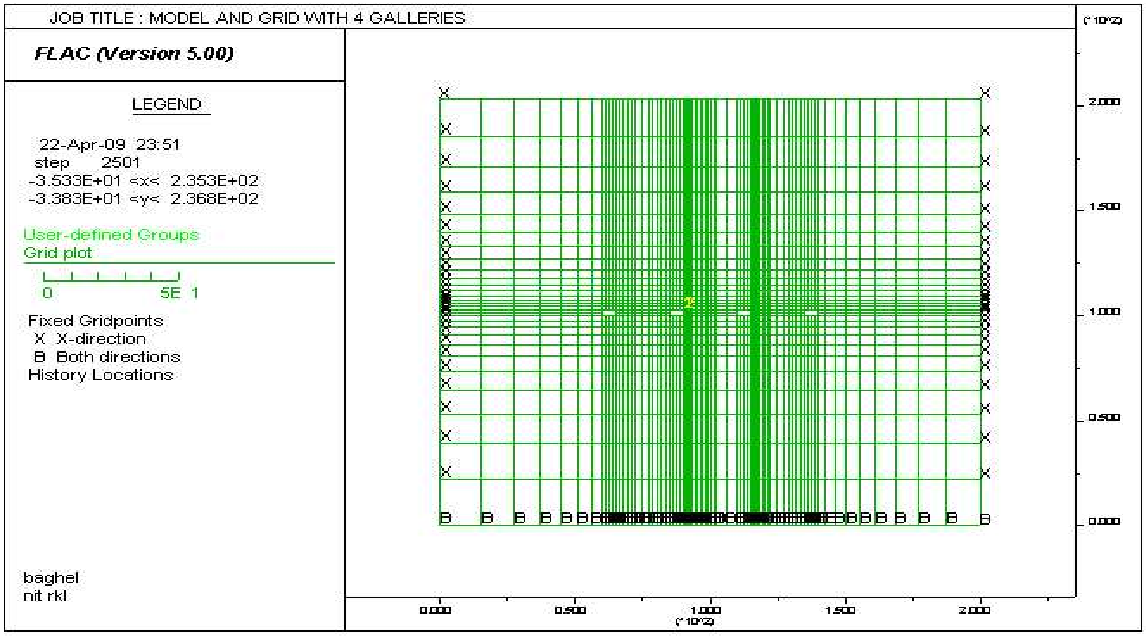
After the extraction of 2 pillar:-

The vertical stress distribution over the goaf and over the rib are found to be $1\times10^6$ Pa and $7\times10^6$ Pa respectively.

After the extraction of a rib:-

The vertical stress distribution over the goaf and at the goaf edges are found to be $2\times10^6$ Pa and $7\times10^6$ Pa respectively.

The stress concentration increases with more extraction, with high stress over rib.



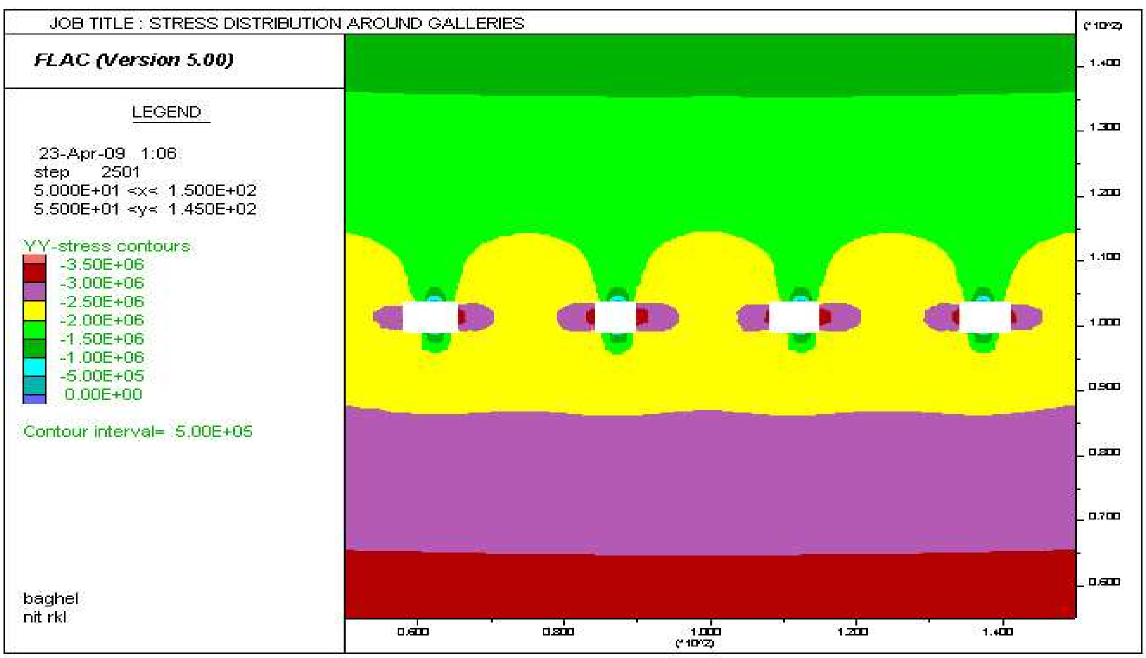
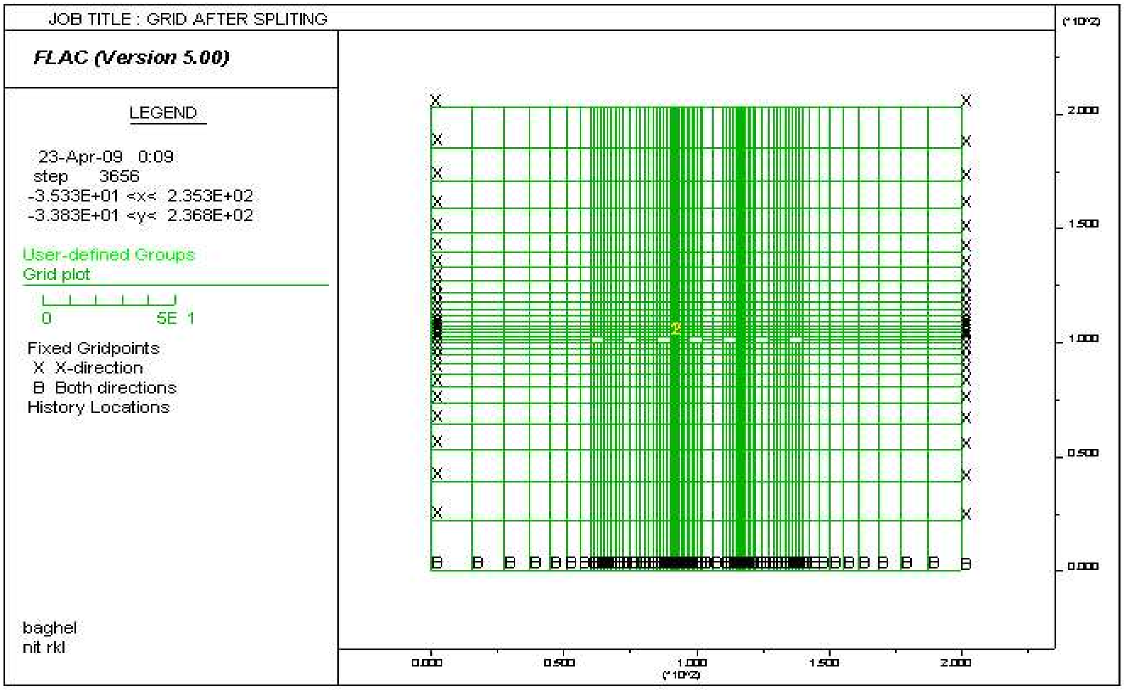




**FIG4. STRESS DISTRIBUTION AROUND GALLERIES**



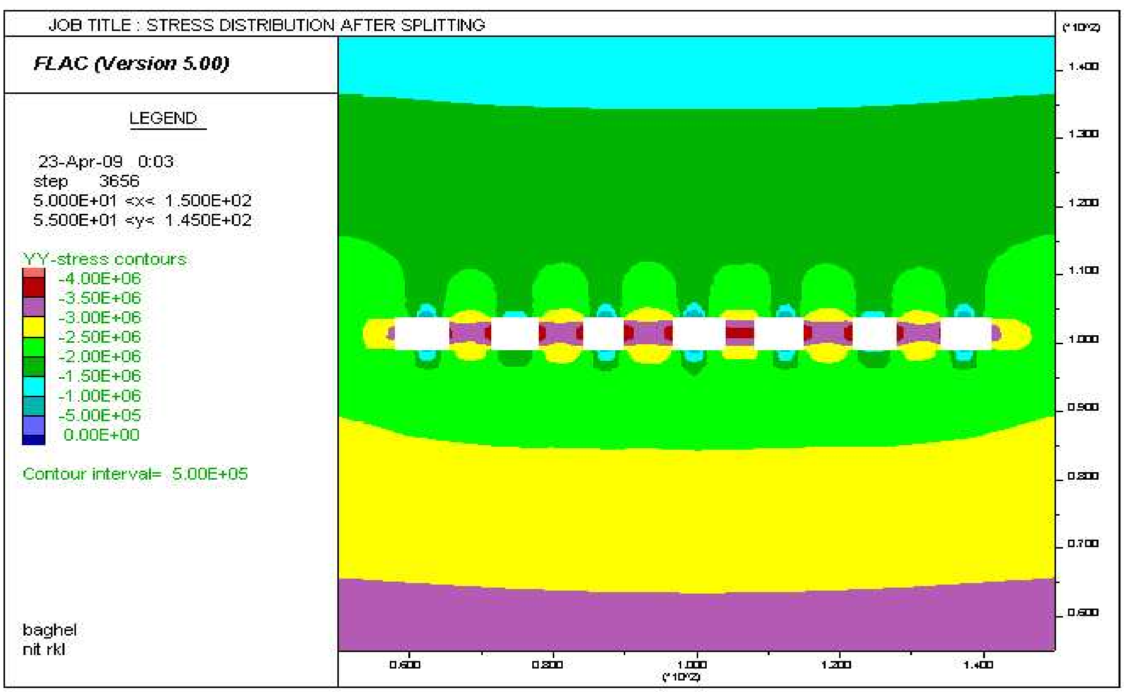
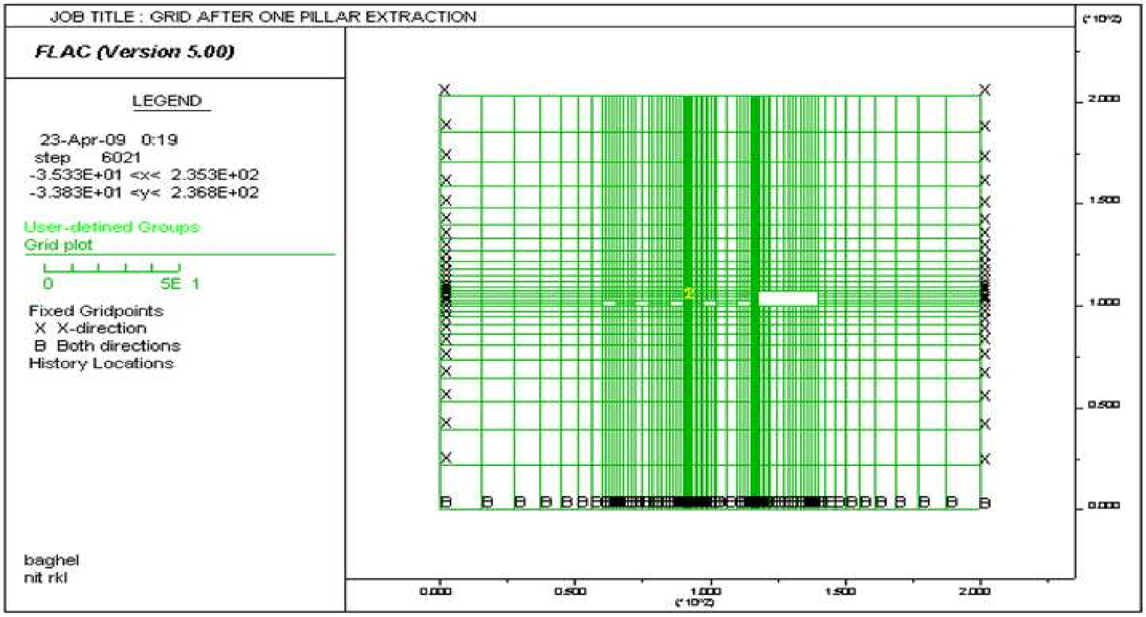




**FIG5. STRESS DISTRIBUTION AFTER SLICING**



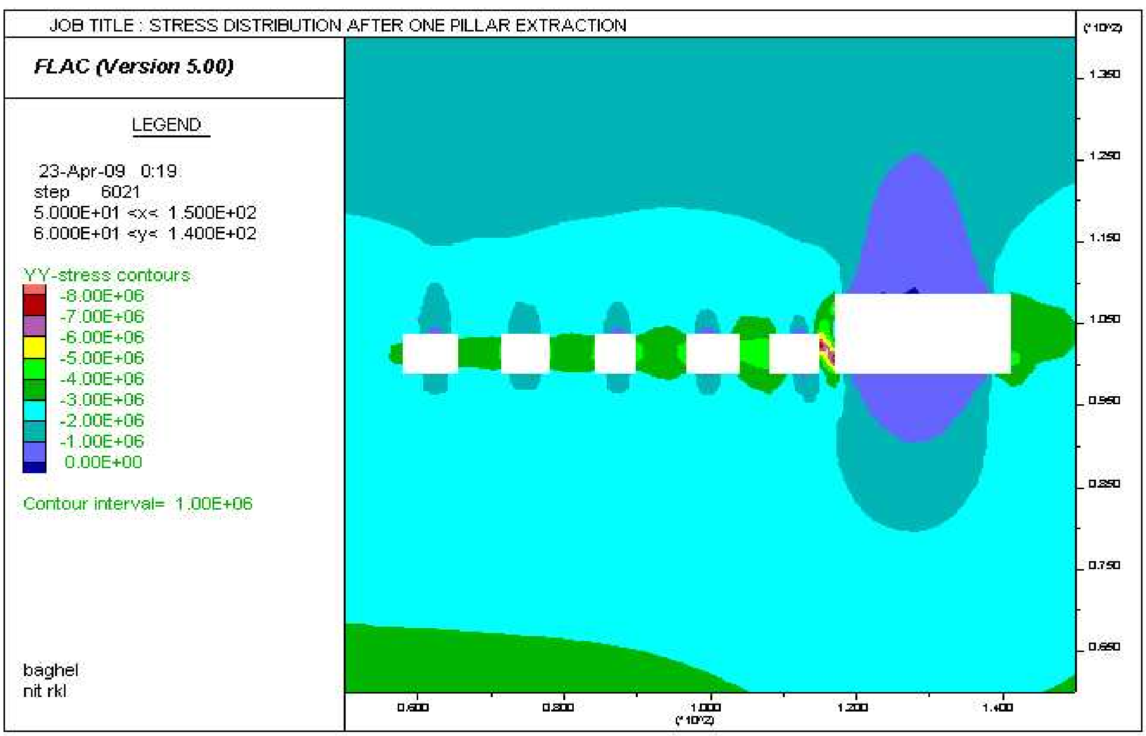
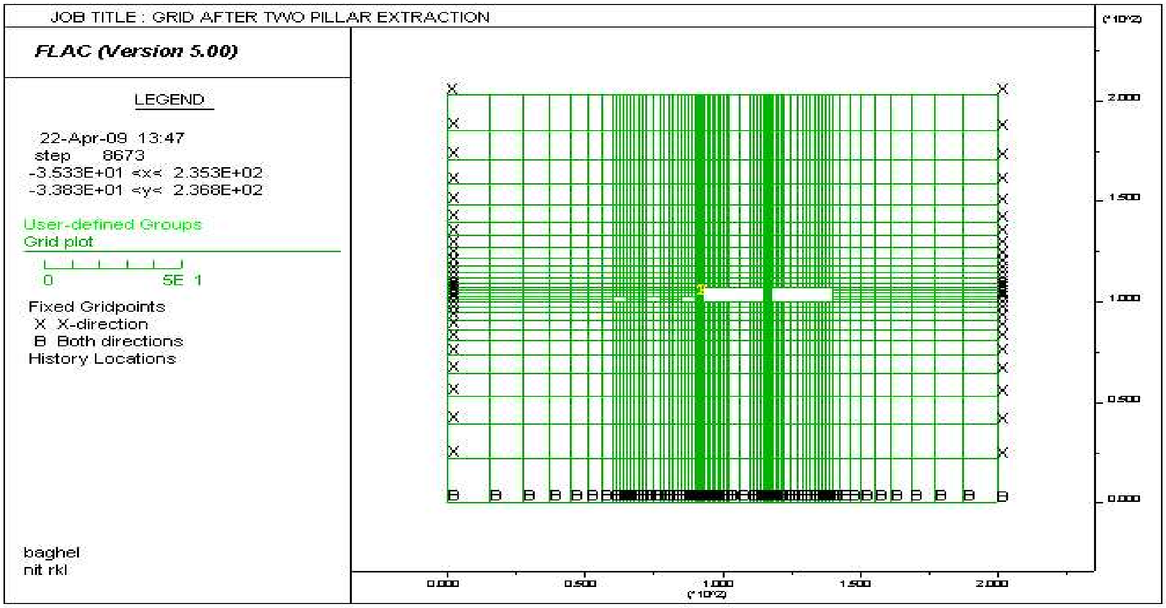




**FIG6. STRESS DISTRIBUTION AFTER EXTRACTION OF ONE PILLAR**



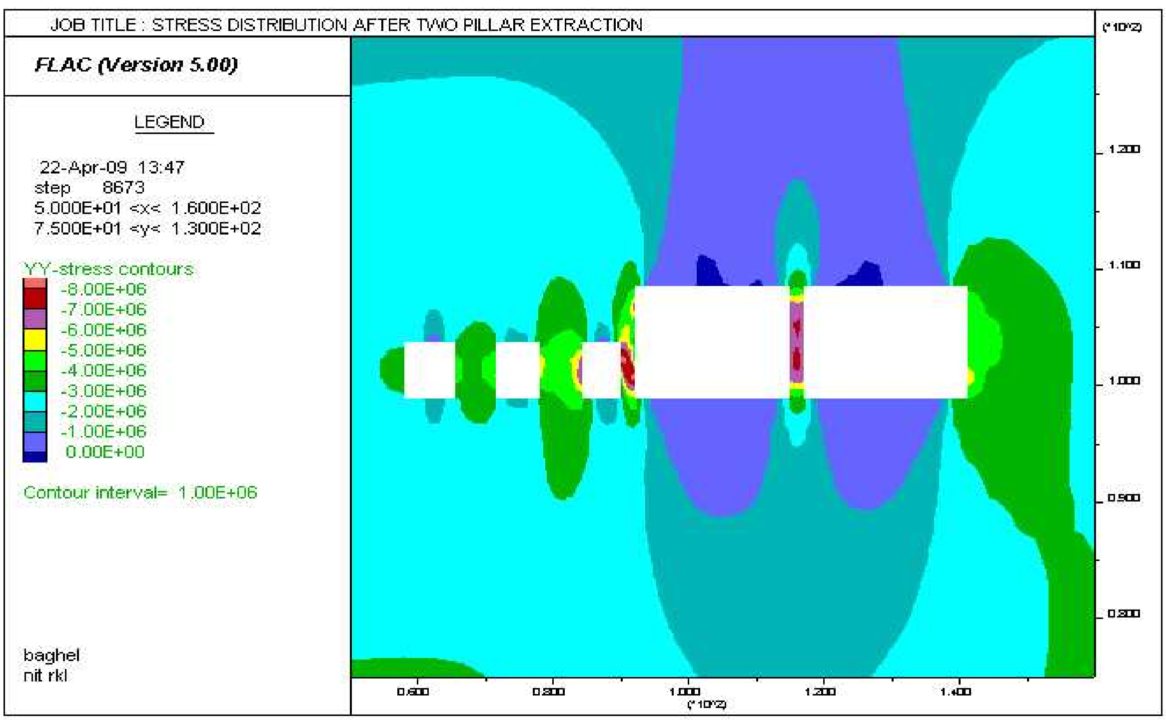
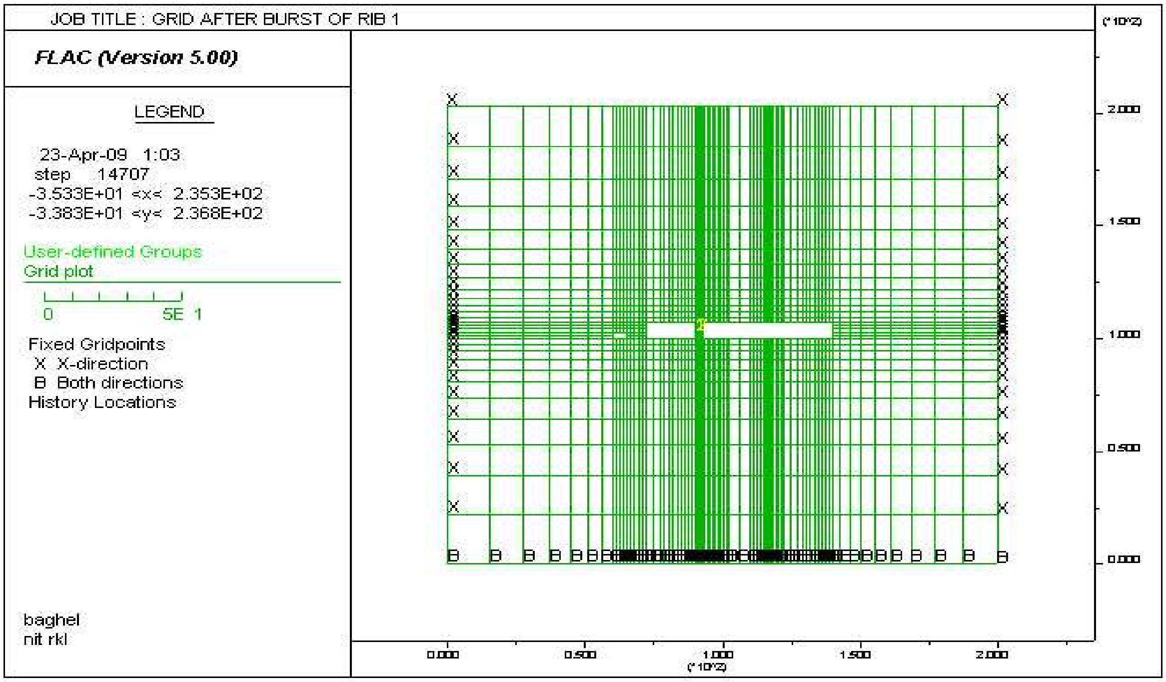




**FIG 7. STRESS DISTRIBUTION AFTER EXTRACTION OF TWO PILLARS**



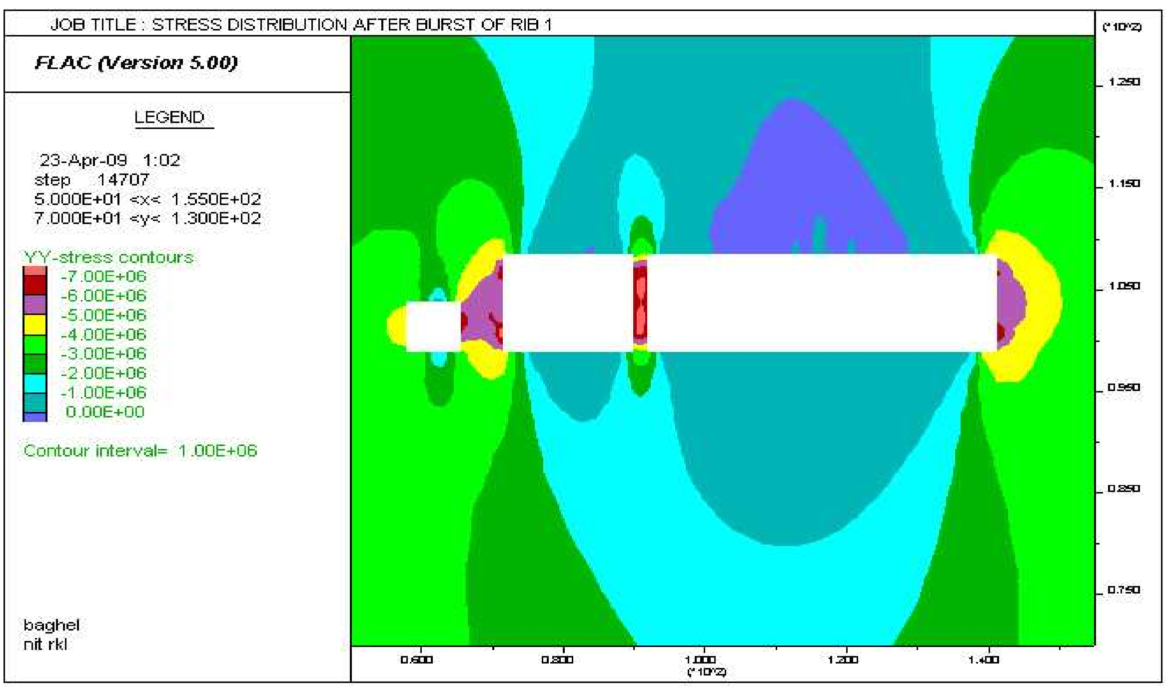




**FIG 8. STRESS DISTRIBUTION AFTER BURST OF A RIB WHERE TWO AND HALF PILLAR HAVE BEEN EXTRACTION**

# 7. RESULTS

From the analysis of stress distribution through numerical model (FLAC5.0) for a depth cover of 95.5m following maximum vertical stress was observed:

- Maximum vertical stress over the pillar during development is about 3 MPa.
- Maximum vertical stress over the stook during development is about 4 Mpa.
- Maximum vertical stress over the rib during development is about 7 Mpa.

# 8. REFERENCES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**(2)** ***S Jayanthu, V Venkateswarlu****, 2008* Strata Behahiour in Development and Depillaring workings supported with Roof Bolt and Cable Bolts, Short Term Course on "**Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines**" **July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 145-153)**

(3) ***Satyendra K Singh****, 2008* Strata Control Techniques and Geotechnical Instrumentation to Coal Mining Industry, Short Term Course on "**Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines**" July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 1-15)

(**4)** ***D. P. Tripathy****, 2008* CMRI-ISM Rock Mass Classification, Short Term Course on "**Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines**" July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 87-97)

**(5)** ***S Jayanthu, Sk. Md. Equeenuddin****, 2008* Geological Factors Contributing to Strata Control Problems in Mines Short Term Course on "**Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines**" July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 120-135)

**(6)** ***S Jayanthu, T N Singh, V Laxminarayana****, 2008* Problems of Underground Coal Mining vis-à-vis Geotechnical Instrumentation for Extraction of Coal, Short Term Course on "**Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines**" July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 136-144)

## 9.1 APPENDIX – I

### PROGRAM CODE FOR SIMULATION OF DEVELOPMENT AND VARIOUS STAGES OF DEPILLARING WORKINGS

```
*PROGRAM DEVELOPED BY ABHINAV S BAGHEL IN GUIDANCE OF PROF S JAYANTHU
*FOR SIMULATION OF VARIOUS STAGES OF PILLAR EXTRACTION IN THICK COAL SEAM

* Seam thickness=7.5m, Pillar size=25m, Depth cover=95.5m
* Gallery size=4.8m X 3m, Width of split=5m ; Rib thickness=2.5m
*PARAMETRIC STUDIES
* Gallery size=4.8m X 3m, Width of split=5m ; Rib thickness=2.5m
GR 78 29
M M
* Floor of the seam -100m
gen 0,0 0,100 60,100 60,0  R .8 .8                          I 1 8   J 1 12
gen 60,0 60,100 64.8,100 64.8,0 R 1 .8                      I 8 12  J 1 12
gen 64.8,0 64.8,100 72.25,100 72.25,0 R 1 .8                I 12 17 J 1 12
gen 72.25,0 72.25,100 77.25,100 77.25,0 R 1 .8              I 17 19 J 1 12
gen 77.25,0 77.25,100 85,100 85,0 R 1 .8                    I 19 24 J 1 12
gen 85,0 85,100 89.8,100 89.8,0 R 1 .8                      I 24 28 J 1 12
gen 89.8,0 89.8,100 92.3,100 92.3,0  R 1 .8                 I 28 33 J 1 12
gen 92.3,0 92.3,100 97.25,100 97.25,0  R 1 .8               I 33 38 J 1 12
gen 97.25,0 97.25,100 102.25,100 102.25,0 R 1 .8            I 38 43 J 1 12
gen 102.25,0 102.25,100 110,100 110,0 R 1 .8                I 43 45 J 1 12
gen 110,0 110,100 114.8,100 114.8,0 R 1 .8                  I 45 49 J 1 12
gen 114.8,0 114.8,100 117.3,100 117.3,0 R 1 .8              I 49 54 J 1 12
gen 117.3,0 117.3,100 122.25,100 122.25,0 R 1 .8            I 54 59 J 1 12
gen 122.25,0 122.25,100 127.25,100 127.25,0 R 1 .8          I 59 61 J 1 12
gen 127.25,0 127.25,100 135,100 135,0 R 1 .8                I 61 66 J 1 12
gen 135,0 135,100 139.8,100 139.8,0 R 1 .8                  I 66 70 J 1 12
gen 139.8,0 139.8,100 200,100 200,0  R 1.2 .8               I 70 79 J 1 12
*
*Coal seam -7.5m
gen 0,100 0,107.5 60,107.5 60,100                          R .8 1 I 1 8    J 12 17
gen 60,100 60,107.5 64.8,107.5 64.8,100                    R 1 1  I 8 12   J 12 17
gen 64.8,100 64.8,107.5 72.25,107.5 72.25,100              R 1 1  I 12 17  J 12 17
gen 72.25,100 72.25,107.5 77.25,107.5 77.25,100            R 1 1  I 17 19  J 12 17
gen 77.25,100 77.25,107.5 85,107.5 85,100                  R 1 1  I 19 24  J 12 17
gen 85,100 85,107.5 89.8,107.5 89.8,100                    R 1 1  I 24 28  J 12 17
gen 89.8,100 89.8,107.5 92.3,107.5 92.3,100                R 1 1  I 28 33  J 12 17
gen 92.3,100 92.3,107.5 97.25,107.5 97.25,100              R 1 1  I 33 38  J 12 17
```

```
gen 97.25,100 97.25,107.5 102.25,107.5 102.25,100            R 1 1  I 38 43  J 12 17
gen 102.25,100 102.25,107.5 110,107.5 110,100                R 1 1  I 43 45  J 12 17
gen 110,100 110,107.5 114.8,107.5 114.8,100                  R 1 1  I 45 49  J 12 17
gen 114.8,100 114.8,107.5 117.3,107.5 117.3,100              R 1 1  I 49 54  J 12 17
gen 117.3,100 117.3,107.5 122.25,107.5 122.25,100            R 1 1  I 54 59  J 12 17
gen 122.25,100 122.25,107.5 127.25,107.5 127.25,100          R 1 1  I 59 61  J 12 17
gen 127.25,100 127.25,107.5 135,107.5 135,100                R 1 1  I 61 66  J 12 17
gen 135,100 135,107.5 139.8,107.5 139.8,100                  R 1 1  I 66 70  J 12 17
gen 139.8,100 139.8,107.5 200,107.5 200,100                  R 1.2 1 I 70 79 J 12 17

*Graphite band  - 10 cm thick
* Sandstone roof-95.5m
gen 0,107.5 0,203 60,203 60,107.5                            R .8 1.2 I 1 8    J 17 30
gen 60,107.5 60,203 64.8,203 64.8,107.5                      R 1 1.2  I 8 12   J 17 30
gen 64.8,107.5 64.8,203 72.25,203 72.25,107.5                R 1 1.2  I 12 17  J 17 30
gen 72.25,107.5 72.25,203 77.25,203 77.25,107.5              R 1 1.2  I 17 19  J 17 30
gen 77.25,107.5 77.25,203 85,203 85,107.5                    R 1 1.2  I 19 24  J 17 30
gen 85,107.5 85,203 89.8,203 89.8,107.5                      R 1 1.2  I 24 28  J 17 30
gen 89.8,107.5 89.8,203 92.3,203 92.3,107.5                  R 1 1.2  I 28 33  J 17 30
gen 92.3,107.5 92.3,203 97.25,203 97.25,107.5                R 1 1.2  I 33 38  J 17 30
gen 97.25,107.5 97.25,203 102.25,203 102.25,107.5            R 1 1.2  I 38 43  J 17 30
gen 102.25,107.5 102.25,203 110,203 110,107.5                R 1 1.2  I 43 45  J 17 30
gen 110,107.50 110,203 114.8,203 114.8,107.5                 R 1 1.2  I 45 49  J 17 30
gen 114.8,107.5 114.8,203 117.3,203 117.3,107.5              R 1 1.2  I 49 54  J 17 30
gen 117.3,107.5 117.3,203 122.25,203 122.25,107.5            R 1 1.2  I 54 59  J 17 30
gen 122.25,107.5 122.25,203 127.25,203 127.25,107.5          R 1 1.2  I 59 61  J 17 30
gen 127.25,107.5 127.25,203 135,203 135,107.5                R 1 1.2  I 61 66  J 17 30
gen 135,107.5 135,203 139.8,203 139.8,107.5                  R 1 1.2  I 66 70  J 17 30
gen 139.8,107.5 139.8,203 200,203 200,107.5                  R 1.2 1.2 I 70 79  J 17 30

PROP S=4.E9  B=6.67E9  D=2300 T=9.E6   C= 12.E6              FRIC=45 I 1 78 J 1 11
PROP S=4.E9  B=6.67E9  D=2300 T=9.E6   C=12.E6               FRIC=45 I 1 78 J 20 29
PROP S=2.2E9 B=3.67E9  D=1427 T=1.86E6 C=1.85E6              FRIC=30 I 1 78 J 12 16
PROP S=1.14E9 B=1.7E9    D=1850 T=.56E6 C=1.1E6              FRIC=35 I 1 78 J 17
PROP S=3.06E9 B=3.9E9  D=1850 T=2.8E6  C=2.1E6               FRIC=35 I 1 78 J 19
PROP S=4.E9   B=6.67E9 D=2300 T=9.E6   C=12.E6               FRIC=45 I 1 78 J 18
SET GRA 9.81
set large
FIX X  I 1
FIX X  J 1
FIX X  I 79
FIX Y  J 1
INI SYY -3.75E6 VAR 0 3.75E6
```

```
INI SXX -4.5E6 VAR 0 0.850E6
HIS NSTEP 10
HIS XDIS I 30 J 14
HIS YDIS I 30 J 14
*Development galleries 4.8m x 3m
HIS UNBAL I 1 J 1
********************OPENING OF GALLERY 1***********
MOD NULL I  8 11 J 12 13
********************OPENING OF GALLERY 2***********
MOD NULL i 24 27 j 12 13
********************OPENING OF GALLERY 3***********
MOD NULL i 45 48 j 12 13
********************OPENING OF GALLERY 4***********
MOD NULL i 66 69 j 12 13
SOLVE
***********************************************
*With developement only* Save as ncdev.sav
***********************************************
Save  ncdev.sav
******Split galleries 5m x 3m
*******************OPENING OF SPLIT 1**********
MOD NULL I 17 18 J 12 13
********************OPENING OF SPLIT 2**********
MOD NULL i 38 42 j 12 13
********************OPENING OF SPLIT 3**********
MOD NULL i 59 60 j 12 13
*******************************************************
*With splitting of pillars Save as ncsplit.sav
*******************************************************
SOLVE
Save  ncsplit.sav
****************************EXTRACTION OF PILLAR 3
MOD NULL I 54 69 J 12 16
**************************************************************
*After extraction of a pillar * Save as ncexp1.sav
**************************************************************
SOLVE
SAVE NCEXP1.SAV
*****************For extraction of two pillars
***************EXTRACTION OF PILLAR 2
MOD NULL I 33 48 J 12 16
SOLVE
SAVE NCEXP2.SAV
```

```
************************************************************
***** FOR EXTRACTION OF 2.5 PILLARS
MOD NULL I 17 27 J 12 16
SOLVE
***** FOR 2.5 PILLARS EXTRACTION - SAVE AS NCEXP25C.SAV
SAVE NCEXP25C.SAV
*****After judicious rob  and burst of rib 1
MOD NULL I 49 53 J 12 16
SOLVE
*********************************************************
SAVE NCEXP25R.SAV
RET
```

## 9.2 APEENDIX – II

## The Coal Mines Regulations (draft), 2006

**124. Support Plan** – (1) The owner, agent or manager of every mine shall formulate a support plan to secure the roof and sides of belowground workplaces, which shall be subject to revision with change in condition, for all workings belowground.

(2) The owner, agent or manager of every mine having workings below ground shall, before commencing any operation frame, with due regard to the engineering classification of strata, local geological conditions, system of work, mechanization, and past experience, and enforce the support plan specifying in relation to each working place the type and specifications of supports and the intervals between:

(i) supports on roadways including places where machinery is used for cutting, conveying or loading;

(ii) each row of props, roof bolts or other supports;

(iii) adjacent props, roof bolts or other supports in the same row;

(iv) last row of supports and the face;

(v) powered supports;

(vi) fore-poles or sprags;

(vii) shields; and

(viii) the pack and the face.

Provided that, in respect of a mine where development operations are already in progress, the support plan shall be framed and enforced within 30 days of the date of coming into force of this regulation.

(3) The manager shall, at least 30 days before the commencement of any operation subject to the provison to sub-regulation (2) submit a copy of the Support Plan to the Regional Inspector who may at any time, by an order in writing, require such modification in the Plan as he may specify therein.

(4) The Manager shall hand over copies of the Support Plan in English as well as in a local language understood by majority of the persons employed in the mine together with illustrative

sketches, to all supervisory officials concerned including the Assistant Manager and Under Manager and shall also post such copies at all conspicuous places in the mine.

(5) The Manager and such supervising officials shall be responsible for securing effective compliance with the provisions of the Support Plan, and no mine or part of a mine shall be worked in contravention thereof.

(6) The support plan shall include inter-alia system of, monitoring of the support performance, measurement of strata behaviour, re-setting of supports, provision of temporary support, replacement of old supports, withdrawal of supports and clearing of falls of ground. The support plan shall also include the implementation strategy of the plan, training and inspection and supervision policies.

(7) The owner, agent or manager shall formulate and implement a code of Standing Orders Specifying:

(a) the system and the organisation for procurement and supply of supports of suitable material,adequate strength and in sufficient quantity where these are required to be readily available for use;

(b) the method of handling including dismantling and assembling where necessary and transportation of the supports from the surface to the face and from the face line to their new site;

(c) the system and the organisation for maintenance and checking of supports, dressing the roof and side erecting, examining and re-tightening of supports and re-erecting dislodged supports, including the use of appropriate tools;

(d) the panel of competent persons for engagement as substitutes in the event of a regular Supports-man or dresser absenting from duty; and

(e) the manner of making all concerned persons such as loaders, dressers, supportsmen, shortfirers, sirdars, overmen and assistant managers including persons empanelled for engagement as substitute supportsman or dresser fully conversant with the support plan and the Codes of Standing Orders under this sub-regulation and under regulation 127 and the nature of work to be performed by each in that behalf.

**125. Use of Powered supports or shields** - The powered supports and shields used in belowground coalmines shall be of a type approved by the Chief Inspector. The approval standards for such supports shall be determined by the Chief Inspector of Mines.

# *SYNOPSIS*

## STUDY OF SUPPORT SYSTEM AND STRATA BEHAVIOUR AROUND UNDERGROUND WORKINGS IN COAL DEPOSITS

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**Supervisor**: Prof. S. Jayanthu

**Abstract:**

The evaluation of the strength characteristics of roof strata during an excavation process while under heavy loads from an overlying rock mass is one of the important factors in strata control of any underground mine.

The work summarizes the results of numerical modeling of the different stages of underground excavation and maximum vertical stress determination in and around pillars, stooks and ribs.

The outcome of the results show that the ultimate vertical stress increases considerably with increase in the depth cover and get concentrated over the area of excavation with high concentration over the pillars, stooks and ribs above the normal stress under the given depth cover.

**Introduction:**

Strata control deals with the adaptation of a system by which we could have a control on the strata movement to a desired level to make our workings safe and extraction of mineral possible. As the future of Indian mining lies in underground workings strata control is of prime importance.

The strata control are to deal with proper management and methodology. In India a good system would result more safer mining atmosphere and high productivity. Geological discontinuities are a prime causative factor in strata-movement problems in underground collieries.

**Objective:**

- To study the stress distribution around development  and depillaring workings in coal mines
- Vertical stress distribution around pillars, stooks and ribs at various stages of pillar extraction studied through numerical models.

**Methodology:**

The working has been modelled by writing a program code in FLAC5.0. The modelling is done in stages involving driving of galleries (development) to form three pillars and then the extraction of these pillars (depillaring) by slicing, then complete extraction to form ribs further followed by the judicious rob and burst of rib. The model is run in each of these stages to get the vertical stress distribution.

**Result:**

From the analysis of stress distribution through numerical model (FLAC5.0) for a depth cover of 95.5m following maximum vertical stress was observed:

- Maximum vertical stress over the pillar during development is about 3 MPa.
- Maximum vertical stress over the stook during development is about 4 Mpa.
- Maximum vertical stress over the rib during development is about 7 Mpa.